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STABILIZED HYDRAULIC MINE BACKFILL: A LITERATURE STUDY

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ABSTRACT

To determine the suitability of the Limestone Injection in Furnace (LIF) by-product as a binding material in a stabilized mine backfill, a two-part study was initiated. The first part was a literature study to collect existing pertinent information on stabilized hydraulic mine backfills. The second part was a laboratory investigation on the properties of mine backfills stabilized with the LIF by-products. This report presents the results and information from the literature study.

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To Dr. T.W. Klym
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**STABILIZED HYDRAULIC MINE BACKFILL:
A LITERATURE STUDY**

1.0 INTRODUCTION

Ontario Hydro is studying a number of processes which may be used to reduce acid gas emission from its coal-fired generating stations in southern Ontario. One process being evaluated at Lakeview TGS involves the direct injection of pulverized limestone (or slaked lime) into the furnace (LIF). A by-product, consisting mainly of limestone (or lime) and flyash, is produced. As part of the evaluation program of this process, studies are being done in Ontario Hydro to investigate the handling/disposal problems of the LIF by-product. Also, utilization options of this by-product are being examined so that the cost of managing a large volume of this material may be minimized.

One of the utilization options of the LIF by-product is to use the material as a binder for the stabilization of hydraulic mine backfills because of the pozzolanic property of the LIF material. In 1987, a research project was initiated in the Civil Research Department to study the feasibility and some of the problems in using the LIF by-product to stabilize mine backfills. The first part of the project was to conduct a literature review on existing information related to the properties and requirements of stabilized mine backfills. Also an understanding of modern mining methods and practices, design of stabilized backfills, etc. was obtained from the literature. The second part of the project was to carry out different types of laboratory tests to measure the properties of various mixes of mine backfill stabilized with the LIF by-products.

This report will only present results from the literature study. The laboratory investigation program will be discussed in a companion report which will be published after this report.

2.0 WHAT IS STABILIZED MINE BACKFILL?

Stabilized mine backfills are similar to weak concrete materials and they consist of an aggregate such as waste rock, mill tailing or sand, a stabilizing agent such as portland cement or alternatives, and water. Backfills with waste rock are generally mixed underground near the stoping area, and then transported to the mined-out stope by a scooptram or mucking machine. Mine backfills with sand or mill tailings as an aggregate are typically batch mixed at the surface and transported hydraulically to the stope area to be filled. The water in the mix reacts with the stabilizing agent to bond the aggregate material, giving the fill cohesion, and an increased internal angle of friction.

There are two distinct types of stabilized backfill, and they are distinguished by their aggregate size. Stabilized rockfill is a backfill with an aggregate of crushed rock of size less than 15 cm, and very little sand or silt sized material. Stabilized hydraulic fill is a backfill with an aggregate generally smaller in size than a coarse sand (less than 1.0 mm), with most of the material in the silt to fine sand range (0.01 mm to 0.3 mm). Each fill type has its own disadvantages and advantages which are discussed below.

Stabilized rockfill has the advantage that it is generally stronger than stabilized hydraulic fill. A comparison between these values for a rockfill and a hydraulic fill can be seen in Table 1. This can mean that a rockfill of adequate strength can be developed with a lower cement cost than a hydraulic fill of similar strength. However, there are also disadvantages. For example, large amounts of waste rock are not always available at mine sites, and expensive quarries may have to be developed. It is also usually more costly to develop a waste pass system to transport the rock underground than it is to install piping for hydraulic fills. These waste passes can also become jammed and have to be cleared. Actual placement of the fill in the stope is also often more expensive for rockfill because of the labour intensive haulage methods used to place the fill. If lower cost methods of rock transport and placement are developed, and if the rock is readily available, the lower cement content and increased strength can make stabilized rockfill a very attractive backfill type (Yu and Counter 1983)/1/.

Currently, stabilized hydraulic backfills are the most popular, even though they can have lower strengths than stabilized rockfills. The reasons that stabilized hydraulic fills are not as strong as stabilized rockfills are quite simple. The hydraulic transport method requires that excessive amounts of water must be added to the fill. This water is necessary to produce a slurry that is pumpable or will flow due to gravity, but it also increases the water/cement (or other stabilizing agent) ratio. When portland cement is used, this high water/cement ratio reduces the effectiveness of the cement and can cause segregation of the cement and aggregate. The much smaller aggregates used in hydraulic fills also lower the strength due to the larger surface area of the particles that must be covered with cement paste, resulting in a thinner, weaker cement bond. The small size of the mill tailings often used in hydraulic fills means that there is also little interaction between the particles, and the result is a lower internal angle

of friction. This loss of strength is, however, balanced by the lower cost of hydraulic fills. The aggregate, in the form of mill tailings, is readily available at the mine site, and the cost of transporting and placing the fill is much lower than stabilized rockfill. These last two properties, abundant aggregate material and low handling cost make stabilized hydraulic fills the most popular backfill type currently in use in the mining industry.

This report focuses only on stabilized hydraulic backfills, specifically those with mill tailings as the aggregate.

3.0 EVOLUTION OF STABILIZED BACKFILL

Fill mining was introduced to Inco mines in the 1920's, and consisted of unstabilized waste rock used in square set, and cut and fill mining. This fill worked reasonably well, but it was difficult and time-consuming to transport and consolidate, so an alternative method was sought. Experiments with smaller fill materials such as sand and gravel gave little improvement over waste rock. Eventually, however, in the mid 1940's, a method of hydraulically placing a fine sand evolved. It was found that by mixing sand with water to produce a pumpable or gravity flowing material, that the fill could be placed more effectively. Thus, hydraulically placed sandfill eventually replaced waste rock as the major filling method at Inco mines and other mining companies.

The advantages of hydraulic placement were numerous. The ease of placement, the ability to get a "tight" fit in the stope, and the greater in-situ density of hydraulic sandfill made it an attractive filling method. This method also provided an environmentally acceptable method of disposing of mill tailings as they were the "sand" in the sandfill. This cut down on large tailings ponds and their associated problems. Thus, hydraulic fills were very practical.

Hydraulic sandfill was an improvement over rockfill, but it still had some drawbacks. If it was to be placed as a vertical face, it needed expensive support. This support was usually provided by some sort of wooden barricade. This was acceptable, but mining close to wooden barricades was difficult, and they were often damaged (Espley et al 1970)/2/. Also, if the sandfill was to be used as the floor material in a cut and fill mining method, wooden "mucking boards" had to be placed on the floor to allow the mucking of broken ore without excessive dilution by the fill material. Underhand cut and fill was not possible with this type of fill, without suspended "mats". The cost of wooden barricades, mucking floors, mats, and their repairs was expensive due to both the materials and labour required. This led to the evolution of stabilized mine backfills.

3.1 Stabilized Hydraulic Fill

Stabilizing mine backfill with portland cement was first used by Inco in the early 1960's, by Noranda in 1964, and other mines in the mid 1950's to early 1960's. It is therefore a new mining technique that has only been in use for about 30 years.

Cemented hydraulic fill* was a great improvement over uncemented fill, as it could stand up vertically and be used as a mucking floor, without causing excessive dilution. Cemented fill has the benefits of easy hydraulic placement plus cohesion, two properties that made it a very attractive fill type. Portland cement was initially used as the main cementing agent, but this has been changing. Portland cement was used at the time of cemented fill development because it was inexpensive and readily available. The chemical and physical properties of a typical portland cement can be seen in Table 2. The behaviour of portland cement had been studied for many years due to its use in concrete, and its dependability made it suitable for use in cemented backfills. These properties; low cost, ease of handling, dependability, and widespread availability, made portland cement an attractive stabilizing agent.

Portland cement has always performed well as a stabilizing agent, but increasing costs initiated research into ways to reduce portland cement content, or to replace it with alternative cementing agents. Research in the 1970's and early 1980's was done on reducing cement content. Studies were done on the effects of aggregate grading and shape, water/cement ratios, and chemical reactions within the fill. Further studies were also done on the design of backfills based on possible failure modes. This research soon reached its full potential in portland cement reduction. Cement content "minimums" were determined, and the only way seen to further reduce costs would be to use alternative stabilizing (cementing) agents.

3.2 Alternative Stabilizing Agents

In order to have an alternative stabilizing agent accepted by the mining industry, it must be proven to perform in a similar manner, or better than portland cement, usually with a reduction in cost. Testing programs must be undertaken to determine the behaviour of fill made with alternative agents. Characteristics such as strength, stiffness, permeability, ease of placement, in-situ performance, and long-term behaviour must be analyzed. It will also be important to show mining companies that an alternative stabilizing agent will be readily available in enough quantity to meet their demand, and can be easily transported to the point of consumption.

Some alternative agents such as natural pozzolans, blast furnace slags, and fly ash have been studied for several years. The majority of tests conducted have focused on the unconfined compressive strength of a fill that is stabilized with one of these materials, as strength is considered the most important parameter in backfill design. In order to complete their acceptance, these alternative agents must also be easily delivered to the mine sites, and in usable quantities. A lower cost, when compared to portland cement, is also usually a requirement for acceptance.

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* A mine backfill stabilized with cement is referred to as a cemented fill in this report.

3.2.1 Natural Pozzolans

Natural pozzolans are those materials found in the earth's crust which can provide silica to react with calcium hydroxide (slaked lime) in the presence of water to form a stable, insoluble, cementitious compound - hydrated calcium silicate. Pozzolanic soils are often mixed with slaked lime ($Ca(OH)_2$) and water to form a cement-like bond. The silicates formed by pozzolanic activity are closely related to those formed by the hydration of portland cement (Thomas et al 1979)/3/. Two examples of the chemical compositions of natural pozzolans can be seen on Table 3.

The reactivity of pozzolans is quite low. It can take several weeks before fill made with a natural pozzolan can be seen to exhibit adequate compressive strength (Manca et al 1983)/4/. If natural pozzolans are to be used, production from the mine must be sequenced such that the fill has adequate time to cure. Another factor to be considered is that the fill must be kept moist during the curing period. These factors may not always be acceptable in a mining operation. Often it may not be possible to sequence the mining operation to allow long curing periods, as the production rate to supply the mill is strictly controlled.

Another shortcoming of natural pozzolans is their availability. They are quite abundant in Italy, where they have been studied the most, but they are scarce in Ontario. For this reason, natural pozzolans are not used as alternative cementing agents in Ontario mines. There are, however, some man-made materials that exhibit pozzolanic behaviour that are used in Ontario mines.

3.2.2 Blast Furnace Slags

The 1980 Canadian production rate of blast furnace slags was about 4.1×10^6 metric tonnes, generally as by-products of nickel, copper, and lead smelting (Douglas and Mainwaring 1985)/5/. These blast furnace slags, especially copper reverbatory furnace slag (CRFS), lead blast furnace slag (LBFS), and quenched copper converter slag (CS) are all man-made materials that exhibit pozzolanic activity. The chemical compositions of these materials can be seen in Table 4 and can be compared with the natural pozzolans and portland cement. The behaviour of blast furnace slag is dependent on certain characteristics and these should be examined before looking at its use in backfills.

It has been found that the pozzolanic behaviour of blast furnace slags is often independent of the chemical composition (silica must be present, but otherwise the other components are not important) but is strongly dependent on the physical condition of the slag (Thomas et al 1979)/3/. The slag must be granulated, and in a "metastable, non-crystalline, glassy condition"; a grind fineness of about 300 m^2/kg provides a good curing rate. Like natural pozzolans, these man-made pozzolans must be kept moist in order to cure properly. This should not be a serious problem in underground mines.

Blast furnace slags have been used in three distinct ways as a stabilizing agent. Firstly, in 1952 in the Horne mine of the Noranda mine group, a fill

composed of 97% granulated slag and 3% pyrrhotite was used. This backfill was analyzed by Nantel and Lecuyer (1983)/6/ as they hoped to use the same type of fill in the Noranda Chadbourne Project. In their study, they examined the unconfined compressive strength, modulus of compression, porosity, heat and SO_2 emissions, and the rate of percolation of water through the fill. They found that because the slag and pyrrohotite they were testing was different from those from the Horne mine, they needed a mix of 90% slag and 10% pyrrohotite tailings to get an adequate fill strength. Mine fills composed of slag and pyrrohotite tailings are rarely used, as pyrrohotite is not widespread, and as shown above, their reactivity can be variable.

The second way in which slags are used as a stabilizing agent is similar to a natural pozzolan. Slag can be mixed with slaked lime ($Ca(OH)_2$) to form a cementing agent for mine backfills. However, the rate of reaction is again very slow. Like natural pozzolans, it can take weeks or months for strength to develop; for this reason slags are not used in this way. If the strength of the fill could be increased in the early stages of curing, then the pozzolanic behaviour of the slag would provide the long term strength. This leads to the third, and most popular, use of blast furnace slags in the mining industry.

If the slags are used in conjunction with normal portland cement, the cement will provide some initial strength, while the slag provides the long-term strength. This line of thinking has lead to the partial replacement of portland cement with blast furnace slags. The combination of these two materials actually causes complementary reactions. The hydration of tri-calcium silicate in the portland cement is accelerated by the addition of the slag, and the slaked lime produced by the hydrolosis of the portland cement reacts with the slag. This results in a mine backfill that exhibits a higher strength than one with portland cement alone (Manca et al 1983)/4/. For example, Thomas et al (1979)/3/ showed that when using portland cement, the addition of 2 to 16% blast furnace slag resulted in increased compressive strength, especially in the long term. Slag mixed with portland cement is being used at many mines in Ontario.

3.2.3 Fly Ash

Fly ash is another man-made material that exhibits pozzolanic behaviour. It too is produced in large quantities each year by industrial processes, such as coal burning. The chemical compositions of some typical fly ash types can be seen on Table 5, and are similar to a natural pozzolan or slag. It is important to note that fly ash can have various compositions, due to the different types of coal burned. Only those from lignite coals are considered as suitable cement substitutes, and those from bituminous coals are deemed unsuitable (Coates and Emery 1970), (Chan 1984)/7,8/. The reason appears to be that the lignite fly ash contains more free lime than the bituminous fly ash. There may also be other physical differences between the two types of fly ash that control their reactivity in a way similar to blast furnace slags. If a bituminous fly ash was mixed with a material with available free lime, it too may prove to be a suitable cementitious material.

Studies/4,8/ have shown that partial replacement of portland cement with fly ash in a cemented backfill has a beneficial effect on strength. For example, Chan/8/ showed that a mine backfill with a ratio of mill tailings to portland cement to fly ash of 94:4:2 (tailing : cement : fly ash) resulted in a backfill approximately 25% stronger than a mix with a ratio of 94:6:0. Also a mix with a ratio of 94:3:3 gave a fill material that was about 50% stronger than the mix with a ratio of 97:3:0. These are promising results that show that lignite fly ash may be added to cemented mine backfills to increase strength.

The mechanism and reasons for strength increase within mixtures containing portland cement and fly ash are similar to those of portland cement and slag mixtures. Fly ash is a pozzolan which requires $Ca(OH)_2$ to harden, and this is supplied by the hydration of the portland cement. Alone they have low strengths, but together their effects are beneficial and result in a higher strength backfill. Recent test results (Yu and Counter 1988)/9/ have shown that fly ash is capable of replacing 60 percent of the portland cement in fills when a three-month curing period is available, and 33 percent when a one-month curing period is available.

4.0 PROPERTIES OF STABILIZED BACKFILL

This section of the report examines those properties of stabilized backfills that are important to the design and performance of the fill. The properties described and the factors affecting these properties are generally for fills containing portland cement, a backfill stabilized with other agents may or may not exhibit the same properties. In this report, a mine backfill stabilized with cement is referred to as a cemented backfill.

In the text Fill Technology in Underground Metalliferous Mines, Thomas et al (1979)/3/ list what they consider to be the nine most important fill properties. These properties are:

1) Relative Density
2) Void Ratio
3) Moisture Content
4) Particle Size Analysis
5) Consolidation Characteristics
6) Compressive Strength
7) Elastic Modulus
8) Internal Angle of Friction and Cohesion
9) Percolation Rate

While all of these properties deserve consideration, many are inter-related and the above list can be modified. This modified list contains five properties that are important to the design and performance of a stabilized backfill, and can be easily studied using standard rock and soil mechanics laboratory procedures. These properties are:

1) Relative Density
2) Particle Size Analysis
3) Compressive Strength
4) Elastic Modulus
5) Percolation Rate, Permeability

The importance of each of these properties will be examined, and the effect of each on the performance of the fill will be discussed.

4.1 Relative Density

The relative density (γ) of the backfill is defined as the ratio of the mass of a sample of fill, to the mass of an equal volume of water. The value of γ is easily determined in the laboratory by following the procedure outlined by Thomas et al (1979)/3/, and it is a good indicator of the strength of the fill.

Mine backfills are generally designed to support their own weight. This "weight-requiring-support" is found by multiplying the density of the fill by some factors related to the shape of the stope, and the anticipated mode of failure.

The relative density of a fill in-situ is generally greater than the value determined in the laboratory. When the backfill is poured into the stope, the dead weight of the fill itself causes it to become very dense. This increasing density generally leads to an increase in strength. The strength increase is likely due to the closeness of the individual fill grains, increasing the internal angle of friction. This problem of a variation between the properties of the fill found in the laboratory, and those in-situ is difficult to resolve. It is impractical to try to match, exactly, the in-situ behaviour of fill in the laboratory. Generally, it has been accepted that laboratory tests done on cemented backfills can be classified as being similar to "index" tests. That is, they give values that can be used to compare different fills, but they do not model the exact behaviour of the fill.

4.2 Particle Size Analysis

The typical output from a mill consists of tailings that range in size from a medium sand to a clay-sized material. The only way to alter the mill tailings is to increase or decrease a certain size fraction, and to see if this affects the behaviour of the fill in any way. Generally, it has been found that mill tailings must be "de-slimed" to improve their performance (Weaver and Luka 1970)/10/. De-sliming is the removal, or reduction of the minus 0.02 mm fraction of the mill tailings. The term "classified" is also used to describe de-slimed tailings, and an example showing a classified and unclassified tailings can be seen in Figure 1. A reduction in the percentage of fines results in a more uniformly graded fill that will have greater permeability.

The effects of particle size on strength have also been studied. Testing done by Thomas et al (1979)/3/ showed that the strength of a cemented back-

fill increased as "fines" were added. This strength increase can be seen in Figure 2, and the difference in the fill grading before and after fines were added can be seen on Table 6. Another study (Manca et al 1983)/4/ showed that a cemented fill made with a "standard" sand exhibited a higher strength than cemented fill made with a tailings sand. These results seemed contradictory. One reason could be that fills of different gradings were used in these two separate studies.

A uniformly-graded material is one that has all particles within a narrow size range, while a well-graded material is one that has a wide range of particle sizes. Somewhere in between a uniformly and well graded material is one that has an optimum grading. This optimum grading is the point where the particles are best able to densify due to their various sizes, resulting in a low void ratio and allowing a minimum amount of cement to fill the voids. Uniform materials will have large voids that will have to be filled with cement paste, and very well graded materials may have particles so tightly packed that cement cannot get into the voids. Both of these materials could have low strength. This concept of an optimum grading has been recognized in concrete technology, and is called the "fineness modulus." This modulus is defined as the sum of the percentages retained on a series of specific sieves, divided by 100. This modulus has not been applied to fill technology, but perhaps a similar modulus should be.

Since there is presently no such thing as a "fineness modulus" for cemented backfills, the only way to find the effects of grading on strength or find the optimum grading is by laboratory testing. Mill tailings samples of various gradings can be tested at a certain cement content, and the optimum chosen. While varying the grading of the fill, it will also be important to make sure that the permeability of the fill is not reduced to an unacceptable value. For this reason, the addition of fines to increase strength is not used in practice.

The shape of the fill particles can also affect strength by altering the internal angle of friction. Particle angularity can increase the internal angle of friction as seen on Table 7. In backfills where mill tailings are used, it would be very hard to control the shape of the particles, but in cemented rockfills, an angular aggregate would be a valuable asset to help increase the strength.

4.3 Compressive Strength

The compressive strength of a stabilized mine backfill is by far the most important of all the properties. This strength is generally determined by uniaxial compressive testing of cylindrical backfill specimens that are mixed, cast, and moist-cured in the laboratory. The development of strength in a stabilized backfill is a very complex topic to study. The strength of a cemented fill is dependent on numerous variables: the water/cement ratio, curing time, curing temperature, grain size, porosity, density, water content, and the chemistry of the aggregate and mix water. As with the density, the strength of a fill in-situ can be quite different from what is achieved in the laboratory. The factors affecting the strength of a backfill are discussed in the following subsections.

4.3.1 Cement Content, Curing Time, and Water/Cement Ratio

The relationships between strength and many of the variables mentioned above are similar to those found in concrete technology. Compressive strength will increase with increasing cement content and curing time as seen in Figure 3, and increase with decreasing water/cement ratios as seen in Figure 4. When considering changes to cement content, curing time or water/cement ratios, it is important to know the limits of each to improving strength.

The lower limit of cement content is that which is just adequate to achieve the desired strength. It is this lower limit that fill designers strive to achieve in order to keep costs low. In very general terms, a cemented backfill placed in open stopes has a cement content of about five percent by weight; backfill for cut and fill mining may reach twelve to twenty percent by weight of portland cement for the "stiff" layers. Any increase in cement content above that required to achieve the desired strength is limited by economics.

The curing time for a cemented backfill also has limits. Production of ore from underground mines is a tightly scheduled operation. Mines must produce ore at a constant rate to supply the mill, or money is lost. Any backfill that is used, must be able to cure to an adequate strength during a certain limited time period, so that production can be on schedule. This time period may vary from a month to several months depending on the size of the mine and mining sequence used. In practice, it is common to use a 28-day curing period when testing backfill specimens for the "design" compressive strength.

The water/cement ratio of a cemented fill is restricted by the flow requirements of the fill. To be able to transport a cemented fill hydraulically, it must be mixed with water at between 60 and 75 weight percent solids. This can mean water/cement ratios of between 1 and 10, whereas a value of 0.4 to 0.55 is usually recommended in concrete mixing. At the present time, however, hydraulic transport is the most economical method to place cemented fill. Applications of concrete pumping technology and pneumatically conveyed fill have been studied (Landriault and Goard 1987)/11/ but have not yet resulted in any feasible improvement over hydraulic placement.

4.3.2 Curing Temperature

Curing temperature is a factor that would affect compressive strength. Compressive strength increases with temperature, until an optimum temperature of about 38°C is reached. Above 38°C, compressive strength, especially in the long term, can be seriously affected. This effect was noticed by Thomas et al (1979)/3/ and is seen in Figure 5.

4.3.3 Grain Size

The effects of aggregate grain size and the shape have been mentioned previously in subsection 4.2. In very general terms, there appears to be an optimum size, grading, and shape for cemented fill aggregates that will

improve strength. There is very little that can be done to alter the size and shape of the mill tailings used in cemented hydraulic fills. Therefore, it is best to test the strength of specimens prepared with various ranges of particle sizes, and to determine the optimum.

4.3.4 Porosity and Density

As previously mentioned in sub-section 4.1, the porosity and density of a given backfill are related. High density indicates low porosity and higher strength. Porosity is generally related to the pulp density* of the hydraulic slurry mix during placement. Higher values of pulp density of the fill during pouring will result in a fill of lower porosity. This relationship can be seen in Figure 6. Little can be done to control these factors except ensuring that the fill is poured at a maximum pulp density.

4.3.5 Moisture Content

The moisture content of a cemented backfill is also related to the pulp density, as well as the relative humidity at which the backfill is cured. Testing (Mitchell and Wong 1982)/12/ has shown that specimens that are "room cured" (at relative humidity 40 percent) show a higher unconfined compressive strength than those that are humid cured (at relative humidity 98 percent). This effect can be seen in Figure 7. Like density and strength, it is very difficult to determine what the moisture content of the fill will be in-situ. For this reason, laboratory samples are generally cured at a high relative humidity (98 to 100 percent) in order to simulate the worst possible conditions that could exist in the stope. This appears to be an acceptable way to account for the possibility of very wet curing conditions that may exist in a cemented backfill pour of large volume.

4.3.6 Chemistry of Aggregate and Mix Water

Chemical compounds in the aggregates or dissolved in the mix water used to prepare cemented backfills can react with the stabilizing agent, and affect the strength of the fill. These effects can be either beneficial or detrimental.

Certain aggregates tend to cause a decrease in the strength of the fill. Studies (Sallert 1979, and Weaver and Luka 1970)/10,13/ have shown that sulphide-rich tailings can affect the hydration and strengthening of portland cement. Cemented backfills made with mill tailings having a high sulphide content can exhibit a strength decrease with time. This is due to "sulphate attack" on the cement bonds, a problem that also occurs in concrete. Sulphate attack occurs when the sulphides are oxidized and form sulphates. Sulphates are larger than sulphides, and this increased size actually "pulls" the cement bonds apart. If these sulphide-rich materials are avoided or larger, less chemically reactive waste rock aggregates are used, the fill should behave adequately.

* The ratio of the mass of solids to the combined mass of the solids and the mix water.

There are also chemical reactions that can occur that will increase the strength of the fill. Tests (Weaver and Luka 1970)/10/ showed that under certain conditions, the addition of very small amounts of flocculants (used in the flotation stage of the milling operation) can increase both the permeability and the compressive strength of a cemented fill. Also, the use of alkaline water (pH around 10) in a cemented backfill mix can increase the compressive strength. Generally, these two ways of increasing strength are not used because they will have different effects under different conditions, and are not reliable ways to increase strength.

4.3.7 Summary

As it has been shown, the strength of a cemented backfill is a very complex property to study. In order to understand these factors affecting the strength of backfills stabilized with portland cement, extensive laboratory testing has been done over the past thirty years. Even though these factors are better understood, many are still being studied. These studies are constantly improving our understanding of how strength is gained in cemented backfills, but most of these factors are difficult to exploit in order to increase the strength. The most promising way to increase strength in fill made with portland cement (without adding more cement) is to increase the pulp density at which the fill is poured. Using present technology, pulp density increases have reached a maximum. Therefore, the only way to increase strength is to increase cement content, which is very costly. A less expensive way to increase strength may be to use alternative cementing agents.

4.4 Elastic Modulus

The elastic modulus, or Young's Modulus, of a cemented backfill indicates the relationship between loading and deformation. Under a certain amount of strain, a material is only able to carry a stress proportional to its elastic modulus. Cemented mine backfills have an elastic modulus that increases with cement content and any other factor that increases strength. The elastic modulus of cemented mine backfills generally increase as the fill cures, and some specific values can be seen on Table 8. These values indicate a maximum value of about 650 MPa. This value may increase to 1,000 MPa, but this is still only a fraction of the elastic modulus of rock, which may be 70 GPa, or more. Typically, the elastic modulus of a cemented fill is one order of magnitude lower than that of the surrounding rock. This means that the backfill will do very little to support the weight of the surrounding rock.

The fact that the elastic modulus of a cemented fill is low, makes it a suitable support mechanism. The low value makes the fill "soft", and capable of absorbing strain energy quickly. This ability to absorb energy makes the fill safe against rockbursting. Rockbursts occur when "stiff" supports fail, and elastic energy stored in the surrounding rock is released much faster than the support can absorb it. The result is an explosive failure. If the backfill was too stiff, large amounts of strain energy could develop in the surrounding rock, and if the fill failed, the failure would be catastrophic. This mode of failure is similar to testing a stiff rock specimen in a soft

machine. This soft nature of cemented backfills compared to the surrounding rock, makes them behave like a soft rock in a stiff machine. This softness is generally not designed, and is simply a characteristic of cemented backfills. The value of the elastic modulus can and should be determined from compressive strength tests.

4.5 Percolation Rate, Permeability

Next to compressive strength, the permeability of a stabilized mine backfill is the most important property for adequate fill behaviour. A stabilized backfill is generally designed to have a percolation rate of greater than 100 mm per hour. This rate is important to the de-watering of the fill after placement. Low permeability of the fill will allow pools of water to develop on top of the fill as seen in Figure 8. If these pools become excessive, they can cause segregation of the aggregate and the cementing agent as the fill is poured through the water. The depth of these pools is often minimized by installing decant towers in large fill pours. The factors affecting permeability, and the application of decant systems are described below.

The percolation rate of a cemented backfill will decrease with increasing fines content as seen on Table 9. The percolation rate(P) of water through a fill can be estimated using $P = 5{,}000\ D^2_{10}$, where P is in cm per hour, and D_{10} is the effective particle size in mm (Smith and Mitchell 1982)/14/, but this is only a rough estimate. The fine particles in a fill cause the material to densify and create a minimum of pore space. It is this pore space that allows water to flow through the fill, and if it becomes too small, water cannot flow through it. The ratio of the volume of this pore space to the volume of the solids, in a given sample, is called the void ratio. Studies (Thomas et al 1979, Keren and Kainian 1983, and Mitchell and Smith 1981)/3,15,16/ have shown that permeability(K) is related to the void ratio as shown in Figure 9, and percolation rate is related to porosity as shown, in Figure 10. Another way that permeability is decreased is by increasing cement content.

The use of cement to increase the strength of mine backfills has been very successful, but it has caused concerns about the accompanying decrease in permeability. Permeability is a great concern in uncemented fills in order to avoid possible liquifaction of the fill. Fill liquifaction is not a concern in cemented backfills because of their cohesive nature. Decreased permeability, however, can cause large pore pressures to develop, and reduce the "effective" strength of the fill. Cement addition decreases permeability because the cement forms a paste that blocks up pores, and limits water flow. An illustration of how cement addition can decrease permeability is seen on Table 10. Thus, a compromise must be made between compressive strength and permeability. This balance will vary depending upon the mining operation and the wetness of the mine.

Concern that high hydrostatic pressures may develop in areas where cemented backfills block the natural flow of water is unnecessary in most Ontario mines, which are usually quite dry. Also, most rock masses, unless highly fractured, have little water flowing through them. The permeability of a

backfill is therefore not important to let outside water flow through, but to let water flow out of the backfill during and after placement.

A discussion of de-watering of large hydraulic backfills pours, including drain numbers, sizes, locations and types can be found in several papers (Smith and Mitchell 1982, Isaacs and Carter 1983, and Thomas et al 1979)/3,14,17/. These papers discuss the various drainage systems that can be used to limit the development of high pore-water pressures. The pouring rate, and sequence can also be used advantageously to limit pore pressure development. These are generally not considered in the actual design and testing of a fill, but more often considered in the placement of the fill.

In the laboratory, the tests done to determine permeability are quite simple. Constant and falling head permeability tests, similar to those used in soils testing are generally used to determine the percolation rate or permeability of fill specimens. In practice, it is common to develop, in the laboratory, a backfill with a minimum percolation rate of about 100 mm per hour.

5.0 REQUIREMENTS OF A STABILIZED BACKFILL

There are several requirements of a stabilized backfill that must be fulfilled before it can be used in practice. These requirements are related to the support provided by the fill, the method of placement, and the mining method. In general, stabilized hydraulic backfills must be strong, durable, permeable, hydraulically transportable, and readily available to mines. These factors are discussed below.

5.1 Strength

Stabilized mine backfills are not designed to support the weight of the surrounding rock. They are designed as free-standing "space fillers" that will support some broken rock or loose "key blocks" and will provide passive support to the surrounding rock. Stabilized backfills, therefore only require enough strength to support their own weight. This requirement is dependent on the size of the stope area being filled, and can range from less than 0.4 to 7 MPa. Stopes in underground mines can reach heights of up to 100 metres, widths can be 25 metres or more, and the length along the strike of the orebody between pillars can be also 25 metres or more, depending upon the competency of the surrounding rock.

A certain amount of strength is also necessary to allow the backfill to stand up while blasting occurs adjacent to the filled stopes. During secondary mining, or pillar recovery, mining is usually done using an open stoping method such as vertical retreat stoping (VRS). As explained in the section of this report on mining methods, VRS mining is done using "blind", large hole blasting. The shock waves produced by these large blasts might induce failure in the backfill. For this reason a factor of safety of 1.2 to 1.6 is sometimes applied to the strength determined, based on the stope dimensions.

5.2 Durability

Durability of a stabilized backfill is closely related to strength. Durability is best described as the ability to resist abrasion and is a necessary requirement where the fill is used as a mucking surface in cut and fill mining. If the fill was not durable, two possible problems would exist. Firstly, the action of scraping broken ore from the floor of the stope would break apart the floor, and cause dilution of the ore with backfill. Secondly, a valuable dense ore (like gold ore), could settle into the floor during mining, and be unrecoverable. Both of these problems could result in lost revenue. For these reasons, portland cement is added to the backfill at a ratio as high as 1:8 to give the floor durability.

5.3 Permeability

As previously stated in subsection 4.5 of this report, permeability is essential to good backfill performance. A percolation rate of greater than 100 mm/hr is a "standard" requirement to allow the fill to dewater itself during and after pouring. Stabilized backfills generally dewater themselves in three ways as seen in Figure 11. These three mechanisms include decantation, exudation, and percolation.

In large pours of stabilized tailings, decant systems are used to aid in the removal of water buildup due to decantation and exudation (Smith and Mitchell 1982)/14/. These decant systems can consist of flexible drainage pipes covered with burlap, supported vertically on wires up through the pipe centres. They are extended through the fill to a drainage area. Excess water within the fill is dissipated by percolation. This combination of a percolation rate of greater than 100 mm/hr and a decant system is generally adequate to allow the fill to dewater, thus creating little danger of failure.

5.4 Hydraulic Transportability

The hydraulic transport of solids in a slurry mixture with a fluid is a very complex topic to study. Entire books have been written on the hydraulic transport of solids, and the relationships between the densities of the solids and the liquid, the viscosity of the liquid, and the particle sizes and quantity of solids in the slurry. All of these factors interact to give the slurry certain flow characteristics at different velocities and pipe sizes. To investigate all of these factors with relation to the hydraulic transport of cemented backfills is beyond the scope of this report. In mining practice, it is standard to combine all of these factors, so that the only property of the slurry that has to be determined and controlled is the pulp density.

The pulp density(PD) of a slurry is defined as the mass of solids(Ms) divided by the combined mass of the solids and the mix water (Ms+Mw). This gives the relation:

$$PD = \frac{Ms}{Ms + Mw}$$

This equation is commonly used in the mining industry as the components of the fill are quite stable. Pulp densities in the range of 65 to 75 percent solids are commonly used in mining practice (Mitchell and Wong 1982)/12/. There is a critical pulp density for a slurry, where the viscosity of the fill starts to increase rapidly. When preparing fill for transport, it is best to stay below the critical pulp density. An illustration of the density-viscosity phenomenon can be seen in Figure 12.

When the LIF is mixed with water, the reaction between the quicklime (CaO) and the water will cause a decrease in the water content of the mix. Each mole of CaO (56g) will consume one mole of H_2O (18g) to produce one mole of $Ca(OH)_2$ (74g). This reaction will lower the mass of the water and increase the mass of the solids, thus increasing the pulp density, making it more viscous and less able to flow. Some water may also evaporate, and the anhydrite in the LIF will consume water to produce gypsum. The gypsum, however, is more soluble, and should remain dissolved in the mix water. In order to overcome this problem, extra water will have to be added to the backfill mix. The amount of extra water added is dependent on the proportion of quicklime in the LIF. An accurate analysis of the LIF will have to be done to determine this proportion. Hopefully this proportion will be constant for a specific feed material, otherwise many samples of the LIF will have to be analysed on a regular basis if the LIF is to be used in practice.

6.0 MINING METHODS AND THE USE OF STABILIZED BACKFILL

To appreciate why and how a mine backfill is used, it is useful to have an understanding of the mining methods and practices. Modern mining methods reflect the demand for high production rates that are required to show a profit. Nowadays, mines are able to produce ore of lower grade if the production rate is high enough, and mining costs are low enough. Mining has become highly mechanical, and bulk mining methods are now employed. Most ore bodies are tabular, or vein-like in shape, and the stoping methods follow this geometry. The mining methods that are presently used to meet these requirements are described in the following subsections.

6.1 Open Stoping

The term open stoping refers to mining methods in which the ore is removed, and the stope area is left open and generally unsupported over large spans. Support is usually provided by leaving pillars of ore to support the roof. Rock bolting and grouting are also used in some cases. Mining of stopes is generally called primary stoping, and the recovery of the ore pillars, which is common in mining, is termed secondary mining. During secondary mining,

there is extensive use of backfill. There are several types of open stoping, each with its own advantages and disadvantages as well as a basic relation to ore body geometry and ground conditions. Most of the large open stoping methods are "non-entry" mining methods. That is, no miners enter the stoping area. In these cases, blasting is done "blindly" from sublevels and ore is removed through drawpoints.

6.1.1 Room and Pillar Mining

This is one open stoping method in which miners enter the stope area. Mining is carried out in large rooms, with pillars of ore being left to support the roof. This mining method is typically used for sedimentary formations, where the ore appears as large flat or gently dipping (less than 30°) beds, or very wide steeply-dipping orebodies. Generally, large, high production equipment is used in these mines to allow them to be feasible, while leaving a large portion of the ore in the form of pillars.

Support in this type of mine is provided on a global scale by the pillars that are left in the mine, and on a smaller scale by rock bolting and grouting to support loose roof-rock. Because the pillars must be left, primary recovery may only be about 60% or lower. In order to increase this value, backfilling is sometimes used to allow secondary mining, if it is judged economically feasible. A good example of a room and pillar operation would be the salt mines in the London and Goderich areas in south-western Ontario. A typical layout of such a mine can be seen in Figure 13.

6.1.2 Sublevel Stoping

This mining method involves sinking a shaft through or beside a steeply dipping ore body. Several sublevels or subheadings are then driven into the ore body at different elevations. The base of the ore body is developed so that a funnel-shaped base and drawpoints can be made. Radial, or long parallel holes are drilled between the sublevels. These holes are charged and blasted so that the ore is removed in vertical "slices". The broken ore collects in the base of the stope and is removed through the drawpoints, without any miners entering the stope. Also, the hanging wall and foot wall must be competent in order to withstand the shock of the blasts and stand up unsupported when the ore is removed. Figure 14 shows an example of a typical sublevel stoping layout.

Mining by this method still does not give 100% primary ore recovery rates. Primary recovery of the ore is still incomplete because rib and crown pillars and the base developments must be left. In order to complete the mining process, pillar recovery techniques are often used. The mined-out stopes are filled with either a stabilized or loose fill material in order to provide support to the surrounding rock, while the pillars are removed. If economically feasible, stabilized fill is preferred, as it provides better stability and lower ore dilution.

6.1.3 Shrinkage Stoping

This method is similar to sublevel stoping in that a steeply-dipping uniformly shaped orebody is required, and the base is developed in the same way. However, shrinkage stoping is a stope entry mining technique. Drilling and blasting of the ore is done in horizontal slices from the base of the ore body upwards towards the top. Ore is blasted in layers, and miners work on the top of the broken ore when drilling the next layer for blasting. The broken ore is also left in the stope to support the sidewalls, with only enough being removed between blasts to allow room for the men to work. The basic layout can be seen in Figure 15.

Because the ore remains in the stope and provides support for the hangingwall and footwall, they can be less competent. The ore, however, must still be competent as the miners will have to work underneath it. The rate of primary recovery is similar to sublevel stoping, and pillar extraction must be done to remove all of the ore.

6.1.4 Vertical Retreat Stoping

This relatively new mining method started in the 1970's and has become extremely popular. It is a cross between sublevel and shrinkage stoping methods. Sublevels are developed, and the ore is blasted in horizontal slices without miners entering the stope area. As in both other mining methods, a basal drawpoint system must be developed. Once this is completed, a horizontal "overcut" (or sublevel) is then made part way up the dip of the orebody. From this overcut, large diameter boreholes are then drilled down the dip of the orebody. Charges are then suspended in these holes at a selected elevation, and horizontal slices of ore are blasted loose. Ore is left in the stope in order to cushion blast effects and to help support the side walls. Because of this, mining may be done in less competent rock. Pillar recovery is again necessary to get as close as possible to 100% ore recovery, and backfilling with a cemented fill is often used. The basic layout of a stope using the vertical retreat stoping method can be seen in Figure 16.

6.2 Cut and Fill Stoping

Cut and fill mining is generally done in narrow, irregularly-shaped, steeply dipping ore bodies, where the sidewalls are not self-supporting. It is a much more selective method than open stoping because it does not use blind blasting; the ore-country rock boundary is usually seen. In this way, dilution is also much lower, and milling costs will be lower. It is also, however, much more labour intensive, and therefore more expensive than open stoping.

As the name implies, cut and fill mining makes extensive use of backfill. The ore is mined in horizontal slices, with fill being used to replace the mined-out material. Fill is usually placed hydraulically, and can be

stabilized or unstabilized, although stabilized fill is preferred. There are two main methods of cut and fill mining and their application depends on the strength of the ore.

6.2.1 Overhand

This is the most widely used cut and fill method. It is used where the orebody is competent. Mining starts at the base of the orebody, and works its way upwards by mining horizontal slices and replacing them with backfill. This type of mining can be seen in Figure 17.

6.2.2 Underhand

This method is not as popular as the overhand method. In underhand mining, the ore is mined downwards, with fill being placed overhead. It is difficult to keep unstabilized fill overhead in this way, so suspended mats must be used. With stabilized fill a similar mat is necessary until the fill gains strength. This method would be used where the orebody was very incompetent, but it is not used too often. This type of mining can be seen in Figure 18.

6.3 Underground Supports

The idea behind the design and placement of supports in underground openings is to allow the rock to support itself as much as possible. After the rock has reached its maximum support potential, rockbolts, chainlink fencing and grout can then be added to help the rock support itself. As excavations are made underground, the rock will deform. The rock is allowed to deform to the maximum allowable limit, and even slight failure may occur before the supports are added. Determining when to place these supports, and how stiff they should be is difficult, and the problem is seen in Figure 19. In stopes, the rock may be allowed to deform and fail more than would be allowed in the shaft or headings.

In a stope, the rock is allowed to deform, and support itself. This deformation allows the redistribution of stresses within the rock so that it may support itself. However, this stress redistribution can cause rockbursts, blockfalls, and spalling of the wall material, all of which can cause dilution of the ore. In order to stop this dilution, stabilized backfill is placed in the empty stopes, thus supporting the loose material. While holding this loose material in place, the backfill also provides passive pressure to the existing pillars and rock, providing added strength. Stabilized backfills are not intended to support the weight of the country rock.

Since stabilized backfills do not support the weight of the country rock, they do little to lower the stresses in the country rock. The stiffness of a stabilized backfill is only about one tenth or less of the stiffness of the surrounding rock. Since the ability of a material to carry a load is related to its stiffness, mine backfill carries little load. Instead of carrying a load, the backfill gives a passive confining pressure to the surrounding rock, allowing it to carry greater stresses.

The modes of support provided by stabilized backfills are numerous. When all stopes are filled with backfill after mining, the chances and amounts of surface subsidence are very low. Stabilized fill provides confining pressure to pillars and surrounding rock, to increase their strength in the same way the axial strength of rock specimens in triaxial tests can be increased by adding confining pressures. Stabilized backfill also provides support to falling wall rock to limit dilution. Backfill helps to absorb blasting shocks and reduce their effects on country rock. Backfilling of stopes can also stop sudden, large displacements of rock caused by rock bursts or global mine failure. These functions: subsidence control, confining pressure, dilution control, shock absorption, and large displacement control all combine to make stabilized backfill a widely used support method.

7.0 MEASUREMENT OF BACKFILL PROPERTIES

The difference between the behaviour of a stabilized backfill in the laboratory and in-situ is a fact that is accepted in the mining industry. This type of behaviour is inherent in any geotechnical work. In both soil and rock mechanics, it is known that the small specimens tested in the laboratory may not describe the actual behaviour of the entire soil or rock mass in-situ. These variations exist due to lateral or vertical variations in soils and the discontinuities in rocks, two conditions that are difficult to simulate in the laboratory. Similar types of variations can exist in stabilized backfills, and the reasons are discussed below.

When stabilized hydraulic backfills are poured, the method of placement often results in a material that is anisotopic and inhomogeneous. The backfills are transported through pipelines and placed at a certain point(s) in the stope. The fill placed at a certain point tends to flow away toward the lowest parts in the stope, due to gravity. In this way, a close contact between the backfill and the stope walls is achieved. This flow of backfill away from a concentrated placement point can cause the fill particles to become segregated. The larger, heavier particles tend to stay at the pour point, while the smaller, lighter particles tend to "float" away. This may cause particle/stabilizing agent segregation resulting in varying backfill strength and permeability across the stope. Using more pour points or moving the pouring point around the stope could minimize this problem.

Another problem, similar to the one above, may also occur, due to decant water collecting on top of a fill pour. When hydraulic backfills are poured, the excess water can "pond" on top of the fill if adequate drainage is not available. If cemented fill is poured through this ponded water, the larger particles settle faster, while the smaller particles stay suspended. This can result in alternating cement rich and cement deficient layers. These layers can affect strength, and seriously lower vertical permeability. Installation of vertical tower drains usually controls this problem, but some segregation may still occur.

Aside from the actual placement, differences in the behaviour between laboratory samples and the in-situ fill can occur due to fill preparation. Variations in the component proportions of an actual stabilized backfill can

occur due to the large volumes of materials being handled, whereas these proportions are usually strictly controlled in the laboratory. Deviations of plus or minus one to five percent can exist in water content, or the proportion of cement to aggregate. For example, at Kidd Creek Mines, the operating pulp density of the fill can range between 45 and 55 per cent solids (Yu and Counter 1988)/9/. These variations can cause a single, large stabilized fill pour to have different properties throughout.

Another factor, not related to fill preparation or placement, can also cause variations between laboratory and in-situ behaviour. That factor is size. The large size of hydraulic backfill pours (up to 100m in height) causes much of the fill to be heavily loaded and compressed while curing. This will result in a lower porosity and permeability, and increased strength with depth. In general, the porosity will be greater in the laboratory than in-situ (Mitchell & Wong 1982)/12/ and the strength follows the same pattern.

Because of all of the variations between laboratory and in-situ fill conditions and characteristics, the values obtained from laboratory testing are used in empirical models of backfill behaviour. Laboratory testing is favoured over in-situ testing because of its ease of execution, low cost, and indications of how the fill would behave in-situ. Below is a discussion of some of the methods of laboratory and in-situ testing used in stabilized backfill practice.

7.1 Laboratory Testing

Presently, laboratory testing is the most common method of determining the suitability of a stabilized backfill for a mining application. Tests done to determine unconfined compressive strength, permeability, density, porosity, particle sizes, and moisture content. For most of these tests, several different procedures could be followed, and approximately the same data obtained. This can be true for density, porosity, particle sizes, and moisture content, and procedures for these can be referenced in soil mechanics texts, and in other papers/3,20/. Tests for strength and permeability, however may yield different results depending upon the method used. Currently, there are no standard tests used in the mining industry (each company, or even each mine site, may have its own testing procedures). Below is a discussion of some procedures used.

7.1.1 Compressive Strength

In order to determine the unconfined compressive strength of stabilized backfills, standard soil mechanics testing apparatus with constant strain may be used. The size and preparation procedure of the test specimens can affect the results. Variations in the size and preparation procedures are discussed below.

The size of the specimen can vary, but should not affect the strength significantly if the height to diameter ratio of the specimen is about 2 to 2.5. This is a standard ratio used in soil and rock mechanics testing in order to ensure that a uniaxial stress state exists in the specimen. They should also

be cylindrical in shape. Some specimen sizes used in the past have been (height: diameter) 30 cm:15 cm/10/, 20.4 cm: 10.2 cm/6/, 10 cm: 5 cm/20/, and 7.6 cm 3.8 cm/3/. all of these have a height to diameter ratio of 2.

The preparation procedure is the area where testing can vary greatly and these variations can affect the strength data. Generally, a stabilized backfill, at the proposed "pouring pulp density", is poured into rigid PVC tubing, allowed to cure, trimmed, and then tested for strength. It is the curing procedure that generally varies between tests and can cause strength variations. As shown in Figure 20, the humidity during the curing period can affect strength quite dramatically. Some test procedures state that the specimen should be allowed to drain/3/ and then placed in a sealed plastic bag for the curing period, others pour the specimen in mold and then place it in a humid room to cure/20/, and others pour the fill into a mold sealed at one end, and then seal the top end once the tube is filled/6/. It is suggested that the fill be poured into a tube sealed at one end and then placed in a moist curing room (98-100 percent relative humidity). By following this procedure, curing at a high moisture content may occur, and the "worst case" of the strength-moisture content relationship can be determined. Because of the low permeability of cemented fills, it is generally assumed that they will cure at a high humidity and moisture content in-situ. The temperature of the specimens should be kept constant during curing, and as close to the expected mine temperatures as possible.

In general, the following procedure should give good strength data that will be close to those found in-situ. The procedure is as follows:

1) Add fill aggregate and stabilizing agent, and mix for a time equivalent to the "retention time" of the hydraulic transport system that will be used to place it. If this time is not known, a time period of about 15 minutes may be used (Espley et al 1970)/2/.

2) The mixed fill is then poured into rigid PVC tubing (sealed at the base) to a depth of about 3 times the diameter. The tubing may be tapped lightly during filling to ensure the absence of voids at the tube walls (The tubing may also be lightly coated with silicon grease prior to pouring to allow easier removal of the hardened fill after curing.) The filled tubes are then placed in a moist curing room and allowed to cure.

3) Once the desired curing period is reached, the specimens can be extruded from the tubes and both ends may be trimmed to get a 2:1 length to diameter ratio. Platens are added to the specimen and it is placed in the testing machine.

4) The sample is then loaded to failure at a constant strain rate, commonly a servocontrolled loading frame is used. The loads and strain can be recorded throughout the test in order to get the elastic modulus, and the load at failure measured to get the compressive strength of the fill.

5) After failure, the angle of failure of the specimen(β) can be measured, and the internal friction angle (ϕ) can be estimated using the equation:

$\beta = 45 + \phi/2$. The sample should also be collected, weighed, dried overnight and then weighed again. This will give the moisture content of the specimen.

6) Approximately 3 to 4 tests should be conducted at each curing period to ensure the accuracy of the procedure, and uniformity of the samples.

By following this procedure, valuable data about compressive strength, curing period and moisture content can be obtained. The samples may also be tested for porosity and density, and these properties noted.

7.1.2 Permeability

As with strength, standard soil mechanics testing procedures may be followed to determine the permeability of the cemented fill. Likewise, there are also several ways these tests are done by various mining companies. It may be difficult to achieve reproducible permeability measurements due to the nature of flow through materials, variations in materials, and possible obstruction of flow paths during measurements. The two main types of permeability tests are the constant head test and the falling head test.

For each of these tests, the fill specimen length is generally 30 cm, and can vary in diameter from 3 cm/21/ to 5 cm/20/. As well, the height of water on top of the specimen can vary from 7.5 cm to less than 1.5 cm. These variations in specimen size and test conditions can cause variations in the final values obtained. As with strength, it would be best to follow a standard procedure, preferably one similar to that used at a mine site.

An example of a procedure for a permeability test is as follows:

1) Mix fill aggregate and stabilizing agent for about 15 minutes, as discussed in compressive strength test procedure.

2) Pour the fill into a perspex tube, about 5 cm in diameter, and having a stiff wire mesh covered with burlap at the base to a depth of 30 cm. Drain decant water from the top of the specimen to ensure a layer of slimes does not exist. Excess fill can be tested for grain size analysis or specific gravity if desired.

3) Add water to the tube to get a depth of water at the top of the fill of $hw = 0.05\ h$, where h is the depth of the specimen (30 cm for this test). Maintain this water level and temperature for several days, and record volume of water (Q) flowing from the base of the specimen.

4) Upon completion of the test, allow the water at the surface to drain into the specimen ($hw = 0$) then weigh the specimen, dry it overnight, and weigh it again. Assuming saturated conditions, this should give porosity and void ratio, if the specific gravity of the solids materials is known.

A sample test arrangement can be seen in Figure 21 (Mitchell and Smith 1979)/20/.

The above procedure is a general one, and variations may exist. In practice, it is a good idea to record the exact procedure, and any variables related to the test. In this way, comparisons may be made between these tests and others.

7.2 Comments on In-Situ Testing

It is known that variations exist between the properties of a stabilized backfill found in the laboratory and those in-situ. These differences are due to the large size difference between the laboratory specimens and the actual stope size, and due to segregation that may occur while pouring the backfill. For the purpose of designing a backfill, in-situ testing results are seldomly used. In-situ testing, however, gains importance when it is used to monitor fill performance to allow variations in the fill that could improve fill performance, or lower costs.

The complexity of the in-situ monitoring of backfill performance can range from simple visual inspection, to complicated testing involving sophisticated equipment. By looking at the amount or lack of fill material being removed from stoping drawpoints, or examining exposed fill faces, the performance of a stabilized backfill can be qualitatively determined. Quantitative values of fill properties can be found by diamond core drilling, plate load testing, dilatometer, and pressuremeter tests. Discussions of these tests can be found in Scoble et al (1987)/22/.

8.0 SUMMARY AND RECOMMENDATIONS

In this report, it is shown that stabilized hydraulic backfill has been used successfully and economically for the last 30 years by mining companies. In most cases, cement is used as a binding agent for stabilizing the backfill materials. Because of economic considerations, a number of mining companies have investigated the use of other stabilizing agents, such as natural pozzolans, blast furnace slags, fly ash, etc. Because of the pozzolanic property of the LIF material, it may be used potentially as a binder to stabilize hydraulic mine backfill.

There are several important properties of stabilized backfill which are important to the design and performance of the fill. They are: relative density, particle sizes, compressive strength, elastic modulus and percolation rate (permeability). The factors affecting these properties are considered in detail in the report. As well, the requirements of a stabilized backfill, such as strength, durability, permeability and hydraulic transportability are discussed.

Various methods for determining in the laboratory the properties of stabilized mine backfill specimens are explained in the report and this information is useful for the laboratory study of LIF/mine backfill materials.

It is recommended that the laboratory study will focus on the strength and percolation rate (permeability) of the LIF/mine backfill specimens. Also,

the physical and chemical properties of the LIF materials should be determined so that a basic understanding of the pozzolanic reactions between the LIF by-product and the filler aggregate (e.g., sand or mill tailings) can be obtained.

In a second companion report, the results of the laboratory study (Part two of the LIF/mine backfill study) will be presented in the near future.

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TABLE 1

COMPARATIVE IN SITU PROPERTIES OF CEMENTED HYDRAULIC FILL AND CEMENTED ROCK FILL*

Property	Cemented hydraulic fill value	Cemented rock fill value
In situ density kg/m^3	1800	2500
Cohesion MPa	0.21	0.50
Unconfined compressive strength MPa	0.8	2.0
Friction angle degrees	35	35-40
Secant modulus at peak strength MPa	150	280

(From Thomas et al. 1979)/3/

* cement hydraulic fill and cemented rock fill are fill materials stabilized with cement

TABLE 2

CHEMICAL ANALYSIS AND PHYSICAL PROPERTIES OF PORTLAND CEMENT

Chemical analysis (wt%)		Physical tests	
SiO_2	20.94	Normal consistency	24.2
Al_2O_3	4.74	Vicat settling time	
Fe_2O_3	2.19	Initial	105 min
MgO	2.51	Final	210 min
CaO	63.03	Fineness- by 75 m	95.3%
CaO (free)	0.60	- by 45 m	86.9%
K_2O	1.14		
Na_2O	0.36	Blaine fineness	3810 cm^2/g
SO_3	3.11	Compressive strength of mortar cubes	
LOI*	2.15	7-day	28.3 MPa
Insoluble	0.58	28-day	35.1 MPa

(From Douglas and Mainwaring 1985) /5/

LOI* = Loss on Ignition

TABLE 3

CHEMICAL ANALYSIS OF NATURAL POZZOLANS

Chemical Analysis (wt%)	Pozzolan from Bacoli	Pozzolan from Sardinia
SiO_2	57.8	64.0
Al_2O_3	18.0	17.0
Fe_2O_3	2.2	3.9
CaO	3.1	1.4
MgO	1.1	0.3
Na_2O	4.1	2.9
K_2O	7.9	4.7
MnO	-	-
SO_3	-	-
Unburnt C	-	-
LOI*	6.0	4.8

(From Manca et al. 1983)/4/

LOI* = Loss on Ignition

TABLE 4

BLAST FURNACE SLAGS
CHEMICAL COMPOSITIONS OF LBFS, CRFS AND CS

Item	Weight percentage		
	LBFS	CRFS	CS
CaO	20.0	7.9	2.4
MgO	1.37	1.9	0.7
SiO_2	17.6	35.5	27.7
Al_2O_3	4.0	3.6	0.6
Fe	27.4	36.0	42.8
S	1.25	0.9	n.a.

(From Thomas et al. 1979)/3/

TABLE 5

CHEMICAL ANALYSIS OF FLY ASHES/8/

Oxide	Thunder Bay Ash		Lakeview Ash
	from Wires* (wt%)	from Plates* (wt%)	(wt%)
SiO_2	40.9	44.5	42.3
Al_2O_3	19.3	18.9	23.2
Fe_2O_3	4.19	4.89	14.7
CaO	11.8	11.5	2.8
Na_2O	5.93	5.12	0.66
K_2O	0.95	0.81	2.1
MgO	2.95	3.17	0.96
SrO	4.32	4.44	-
TiO_2	0.97	0.92	-
BaO	1.15	1.12	-
P_2O_5	0.62	0.52	0.49
SO_3	10.2	5.23	0.87
LOI	-	-	7.96

* - Wires and plates inside the ash precipitator

TABLE 6

GRAIN SIZE ANALYSIS OF FILL SAMPLES SHOWING SIZE FRACTIONS BEFORE AND AFTER THE ADDITION AND REMOVAL OF FINES

Fraction	Wt. %			
μm	As-received 1	As-received 2	Fines added to 1	Fines removed from 1
+ 425	0.10	0.42	0.09	0.11
300 - 425	0.34	0.40	0.40	0.39
212 - 300	1.83	1.35	1.78	2.01
150 - 212	4.53	7.94	3.48	4.67
106 - 150	20.99	17.87	21.06	23.93
75 - 106	20.82	20.54	19.86	22.08
53 - 75	18.39	17.14	17.92	19.04
41 - 53	3.27	4.54	3.96	3.55
31 - 41	13.01	14.27	11.84	12.20
21 - 31	4.26	7.04	5.25	3.81
14 - 21	2.62	0.02	3.64	1.79
10 - 14	1.93	1.01	2.17	0.97
- 10	7.93	5.22	8.61	5.44

(From Thomas et al. 1979) /3/

TABLE 7

EFFECT OF PARTICLE ANGULARITY AND GRADING AND PACKING DENSITY ON FRICTION ANGLE

Shape and grading	Loose	Dense
Rounded, uniform	30°	37°
Rounded, well graded	34°	40°
Angular, uniform	35°	43°
Angular, well graded	39°	45°

(From Thomas et al. 1979) /3/

TABLE 8

MODULUS OF ELASTICITY FOR CEMENTED FILL

Cement content	Modulus of elasticity	
Wt. %	At 7 days	At 28 days
	MPa	MPa
2 (1)	1.21×10	2.57×10
4 (1)	2.48×10	4.76×10
6	9.10×10	9.03×10
8	1.39×10^2	1.85×10^2
10	1.67×10^2	2.06×10^2
12	2.06×10^2	3.01×10^2
14	3.94×10^2	4.67×10^2
16	5.58×10^2	6.43×10^2

Footnote: No real linear section exists in stress-strain curve. Values are estimates only.
(From Thomas et al. 1979)/3/

TABLE 9

GRAIN SIZE AND PERCOLATION DATA

	D_{10}	D_{15}	D_{50}	D_{85}	P_{calc}
Tailings	0.044	0.055	0.12	0.25	9.7 cm/hr.
Medium Sand	0.2	0.24	0.5	1.9	200 "
Coarse Sand 1	0.65	0.84	1.3	2.3	2000 "
2	0.84	0.84	1.5	3.4	3500 "

Note: The calculated percolation rates are given by $P_{calc} = 5000 D_{10}^2$ cm/hr (all D_{10} values in mm).

(From Smith and Mitchell 1982) /14/

TABLE 10

PERMEABILITY OF CEMENTED HYDRAULIC FILL

Portland cement content	Void Ratio (1)	Permeability, expressed as a percentage of value without cement
Wt. %	-	%
0	0.689	100
4	0.664	25.4
8	0.644	10.1
12	0.646	5.03
16	0.603	2.43

Footnote. 1. Void ratios achieved with constant compactive effort.

(From Thomas et al. 1979)/3/

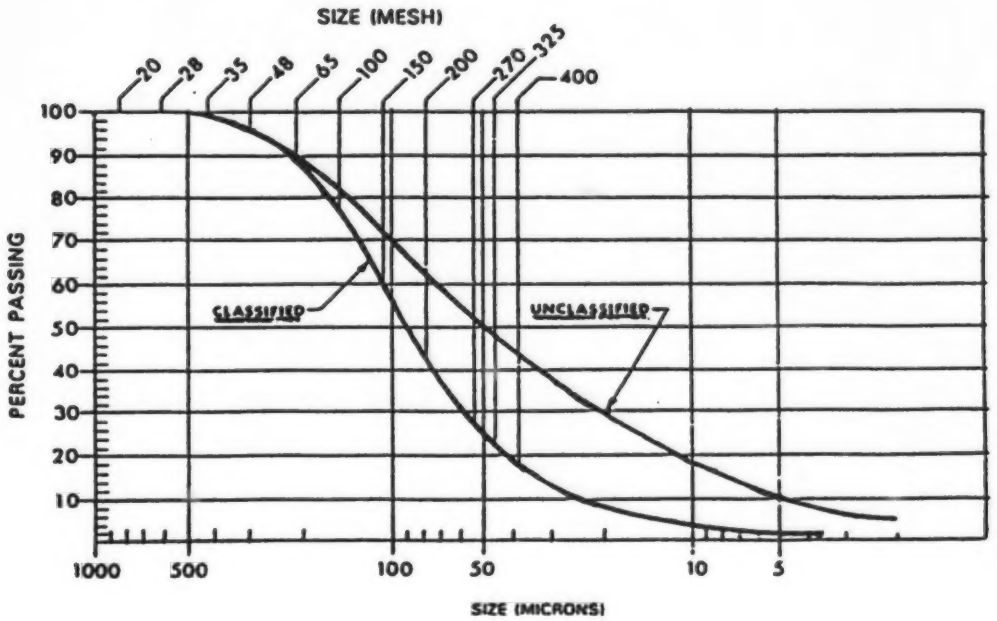
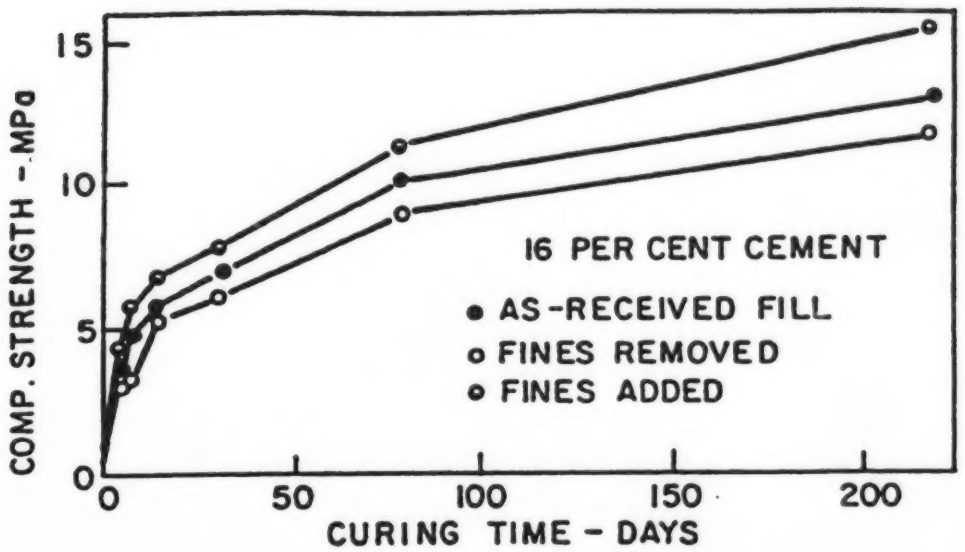


FIGURE 1

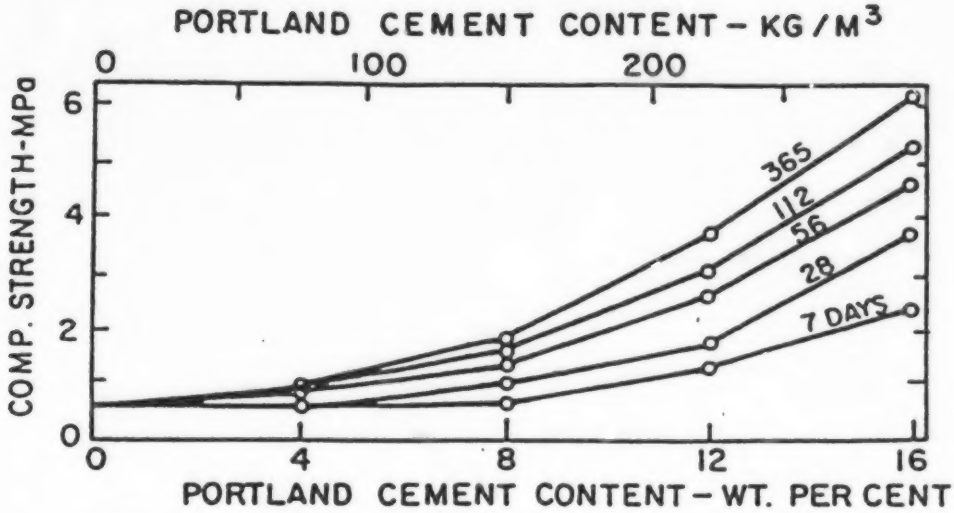
COMPARISON BETWEEN AN UNCLASSIFIED AND
A CLASSIFIED MILL TAILINGS



(From Thomas et al. 1979)

FIGURE 2

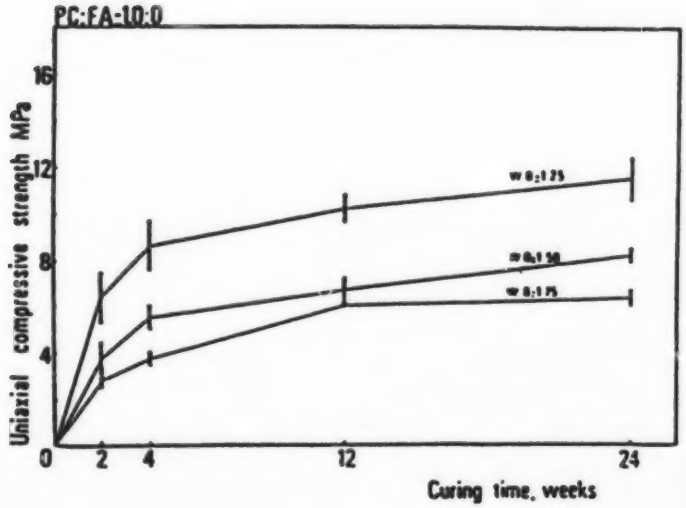
THE EFFECT OF ADDING OR REMOVING FINES ON
CEMENTED BACKFILL STRENGTH FOR A PARTICULAR FILL SAMPLE



(From Thomas et al. 1979)

FIGURE 3

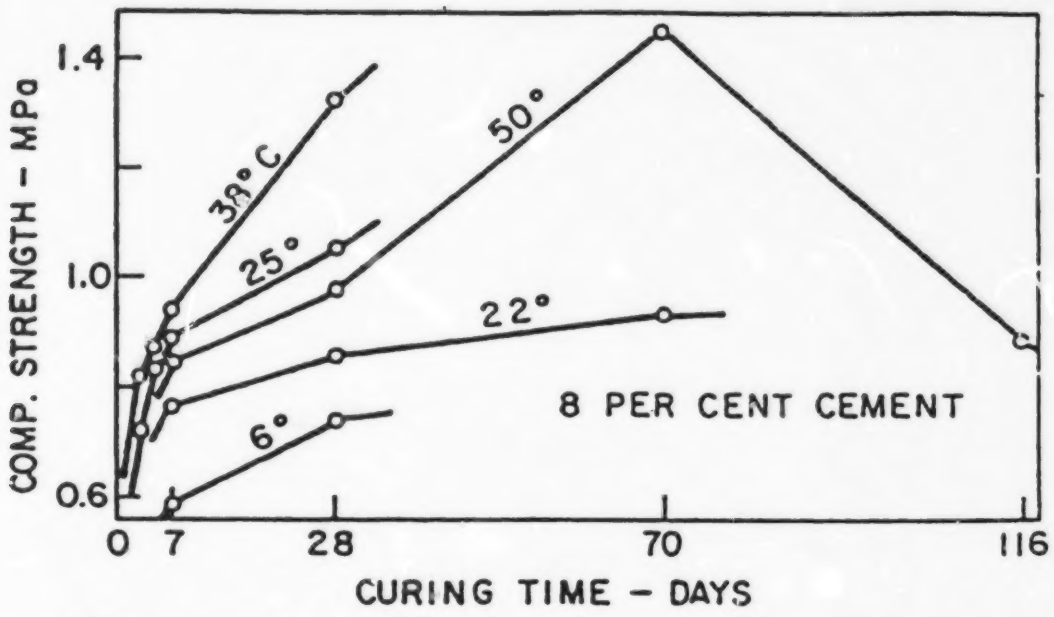
THE EFFECTS OF CEMENT CONTENT AND CURING TIME ON THE STRENGTH OF A CEMENTED FILL



(From Manca et al. 1983)

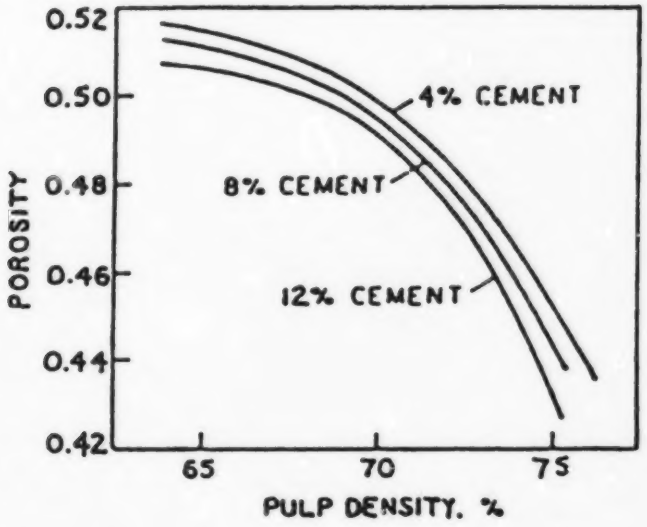
FIGURE 4

EFFECT OF VARIOUS WATER:BINDER (WATER:CEMENT) RATIOS ON BACKFILL STRENGTH



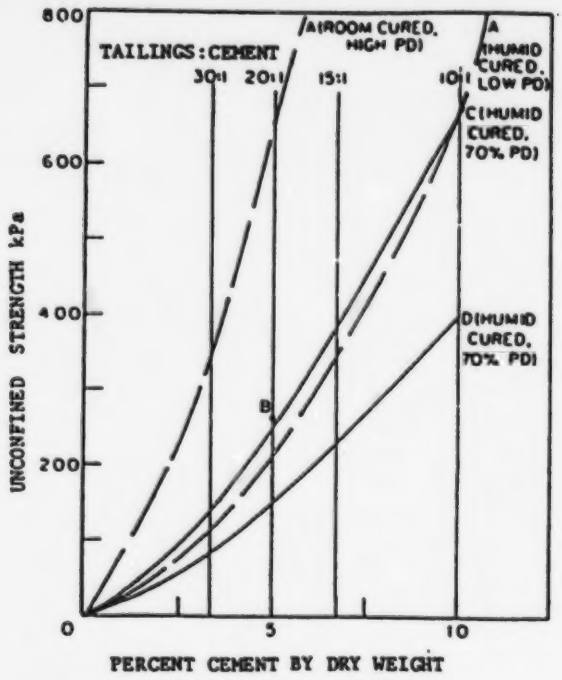
(From Thomas et al. 1979)

FIGURE 5
THE EFFECT OF CURING TEMPERATURE ON FILL STRENGTH



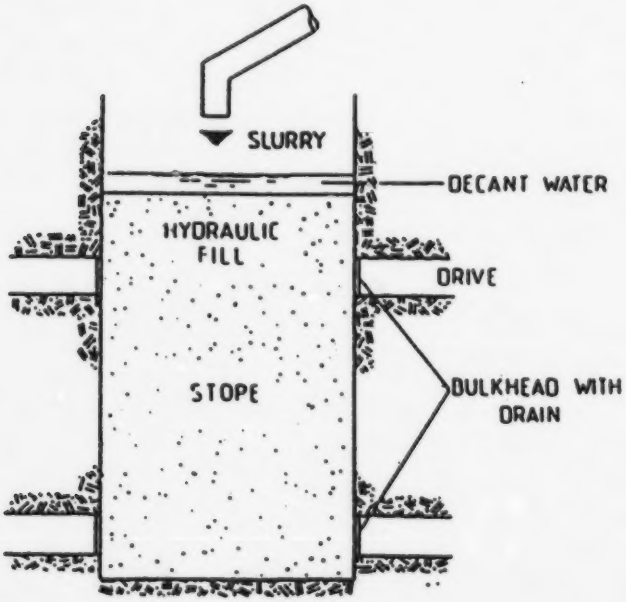
(From Mitchell and Wong 1982)/12/

FIGURE 6
RELATIONSHIP BETWEEN POROSITY AND DENSITY



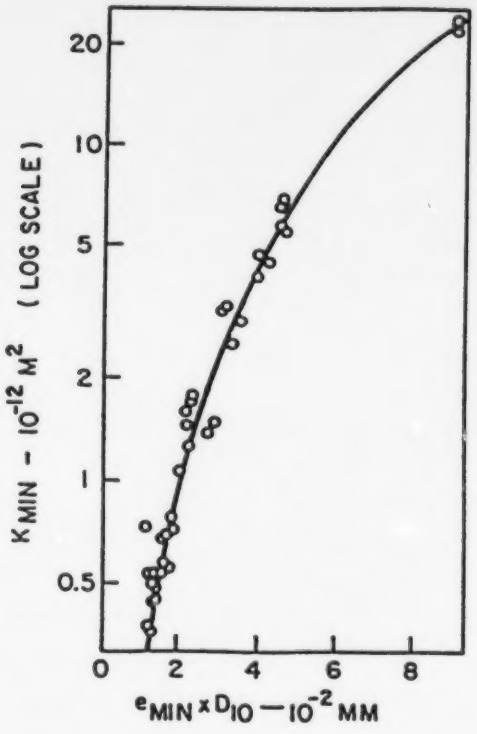
(From Mitchell and Wong 1982)

FIGURE 7
EFFECT OF RELATIVE HUMIDITY DURING CURING ON STRENGTH



(From Isaacs and Carter 1983)

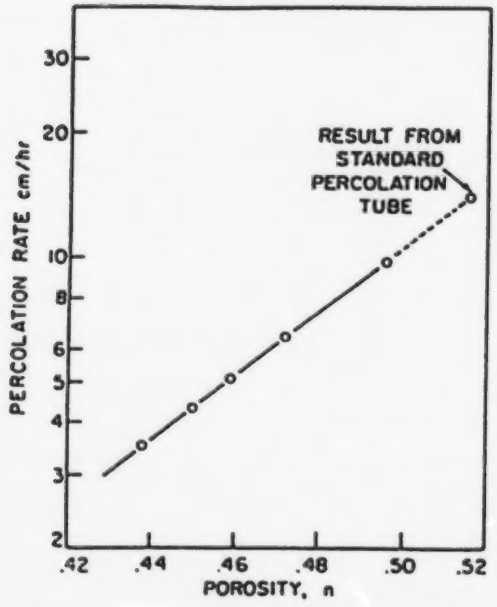
FIGURE 8
ILLUSTRATION OF DECANT WATER CAUSED BY FILL POURING



(From Thomas et al. 1979)

FIGURE 9

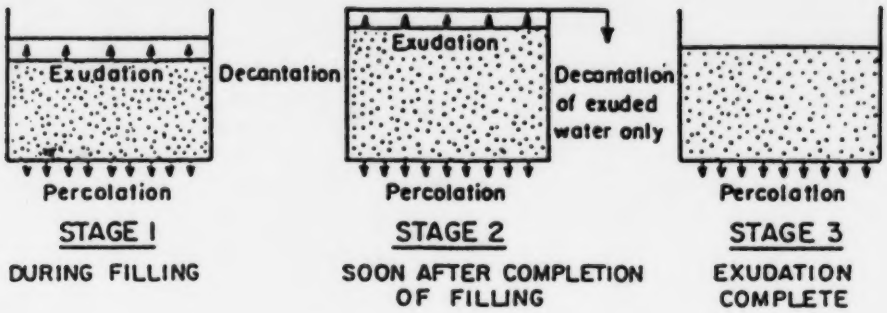
EFFECT OF VOID RATIO (e) AND GRAIN SIZE (D_{10}) ON PERMEABILITY (K)



(From Mitchell and Smith 1981)

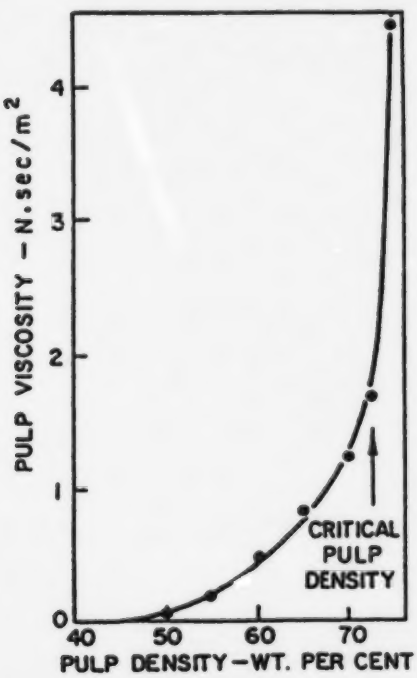
FIGURE 10

EFFECT OF POROSITY ON PERCOLATION RATE



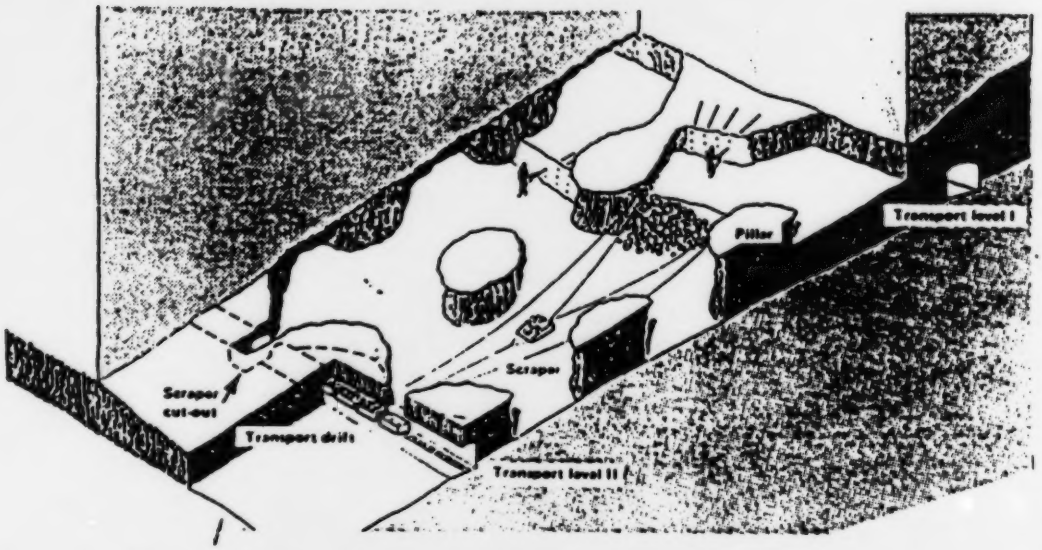
(From Thomas et al. 1979)

FIGURE 11
THE THREE STAGES OF CEMENTED BACKFILL DE-WATERING



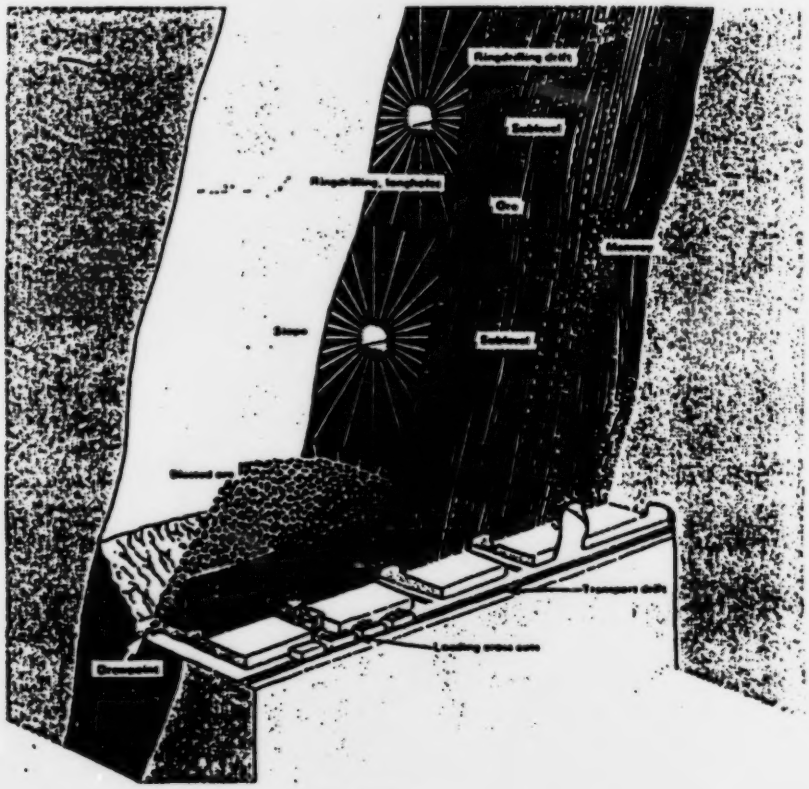
(From Thomas et al. 1979)

FIGURE 12
EFFECT OF PULP DENSITY ON BACKFILL VISCOSITY
(AND THEREFORE BACKFILL FLOWABILITY)



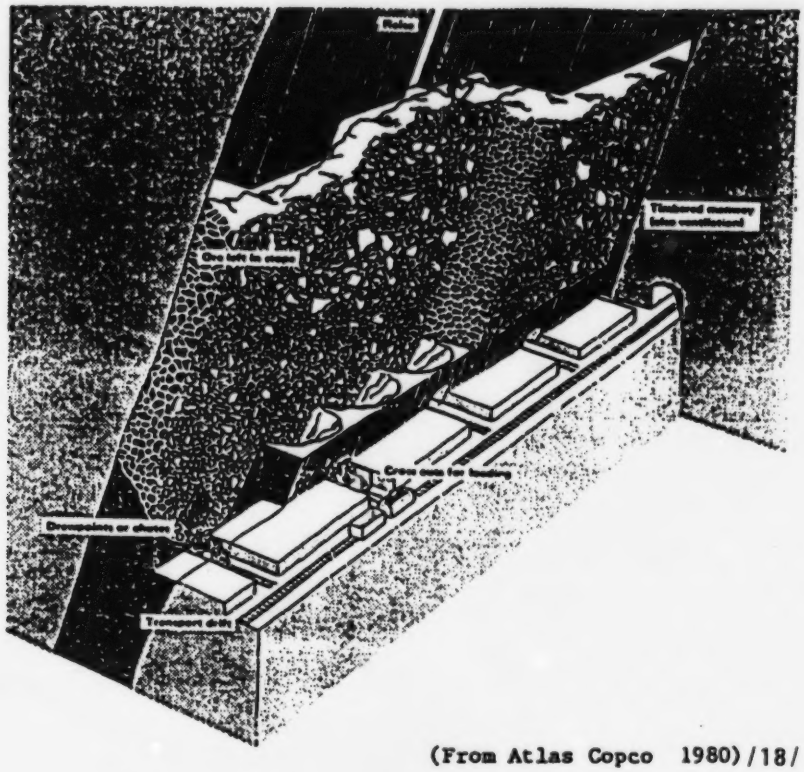
(From Atlas Copco 1980)/18/

FIGURE 13
LAYOUT OF A ROOM AND PILLAR MINE IN AN INCLINED OREBODY



(From Atlas Copco 1980)/18/

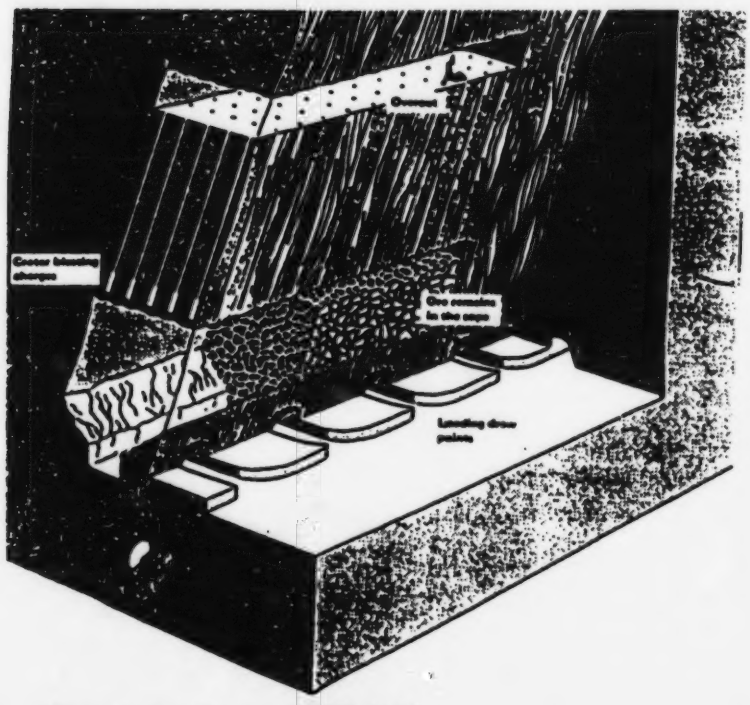
FIGURE 14
LAYOUT OF A SUBLEVEL STOPING OPERATION



(From Atlas Copco 1980) /18/

FIGURE 15

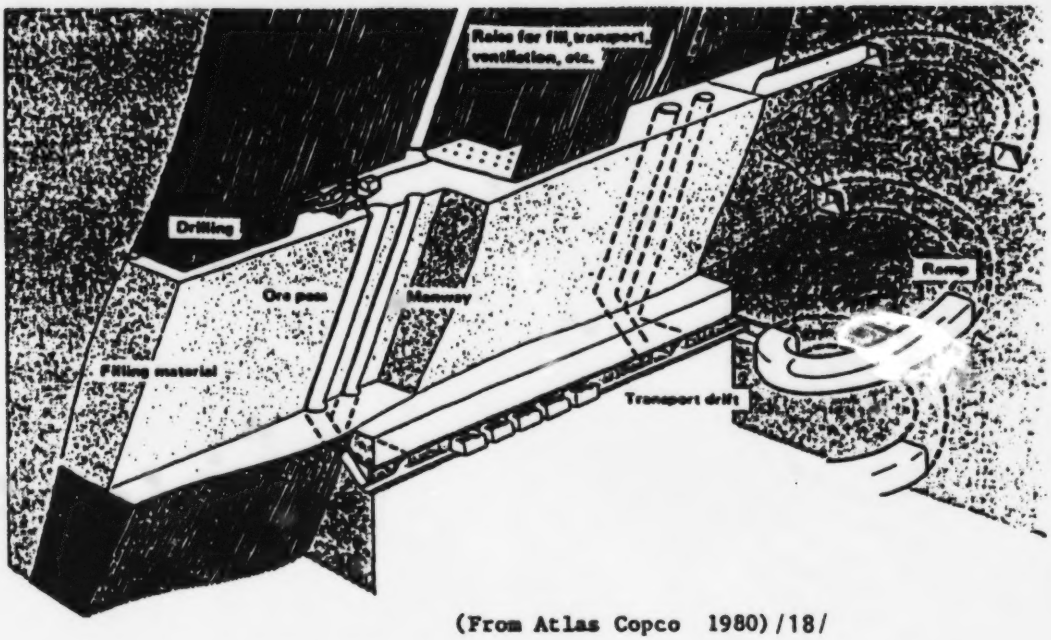
LAYOUT OF A SHRINKAGE STOPING OPERATION



(From Atlas Copco 1980) /18/

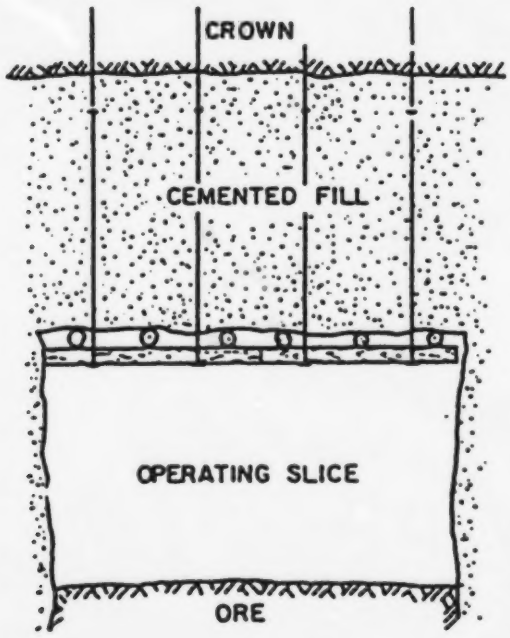
FIGURE 16

LAYOUT OF A VERTICAL RETREAT STOPING OPERATION



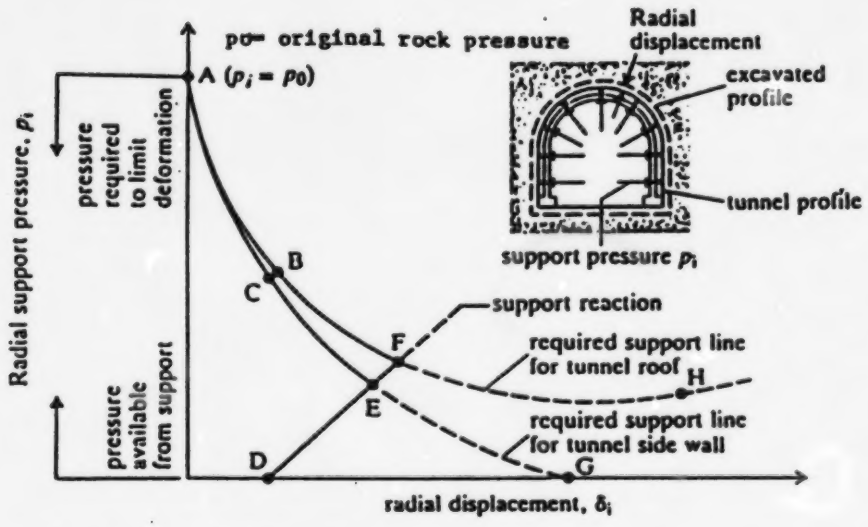
(From Atlas Copco 1980) /18/

FIGURE 17
LAYOUT OF AN OVERHAND CUT AND FILL STOPING OPERATION



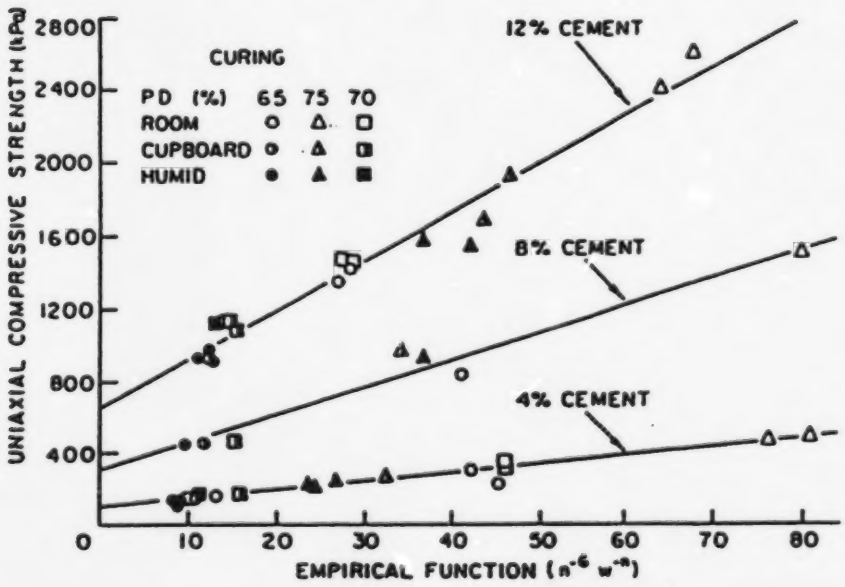
(From Thomas et al. 1979) /3/

FIGURE 18
STOPING ARRANGEMENT IN UNDERHAND CUT AND FILL STOPING



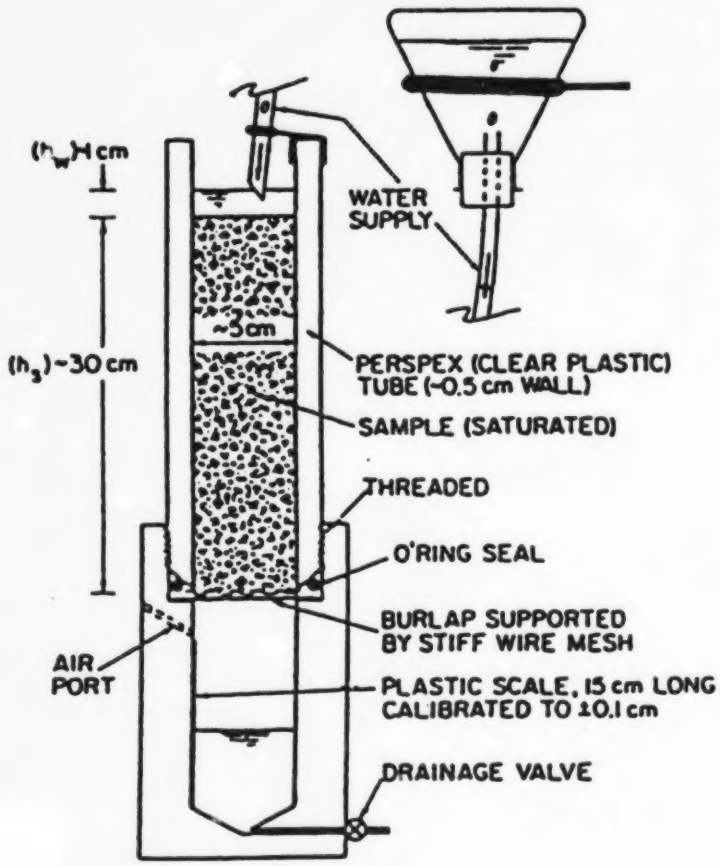
(From Brady and Brown 1985) /19/
Note- Slope of line DE is proportional to support stiffness.

FIGURE 19
EFFECT OF EXCAVATION ON ROCK DEFORMATION AND REQUIRED SUPPORT PRESSURE



(From Mitchell and Wong 1982) /12/

FIGURE 20
EFFECTS OF POURING PULP DENSITY AND CURING CONDITIONS ON CEMENTED BACKFILL STRENGTH



(From Mitchell and Smith 1979)

FIGURE 21
SAMPLE PERCOLATION TEST ARRANGEMENT
FOR CEMENTED BACKFILL